Controlled Recirculation in Deep South African Gold Mines

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ABSTRACT

Despite the many advantages and potential benefits of using controlled recirculation in Deep South African gold mines the strategy has found little favour. Indeed, controlled recirculation of air is at present rarely considered as an option in the planning and implementation of these underground ventilation systems. This paper broadly examines controlled recirculation, the benefits and advantages that could accrue from its use, reasons that may detract from its use and some solutions for addressing shortcomings. A brief historical review of controlled recirculation in the context of the South African gold mining industry is given. The advantages, which could result from its use, are examined in the context of a Deep South African gold mine, by means of computer simulations. Comparative cost appraisals, derived from the simulation results, are also given.

KEYWORDS

Controlled Recirculation, Deep Gold Mine, and Conceptual Model.

INTRODUCTION

The increasing depth of South African gold mines, typically 3,000 m below surface, has led to an escalation in the logistical problems and costs associated with conditioning the thermal environment to the point that new and innovative methods and systems to achieve acceptable conditions must be considered. Two broad strategies with the potential to reduce the impact of increased depth on the thermal environment exist.

1. Developing methods to minimize the heat flow into the ventilation air. The major causes of high heat loads in deep mines are heat flowing from the exposed rock and autocompression of the ventilation air.
2. Conditioning the environment. To achieve this, deep South African gold mines rely upon refrigeration. Typically, chilled water from refrigeration plants is stored in large underground dams and then distributed to the mine workings as required. The primary medium for distribution of cooling within the working areas is the ventilation air.

The effectiveness of both of the above strategies can potentially be improved by the implementation of controlled recirculation. Although many of the inherent advantages of controlled recirculation increase with increased mining depth, the implementation of controlled recirculation on deep South African gold mines has been disappointing. This paper strives to briefly examine the background and determine the impact that controlled recirculation may have on a deep mine. The latter is accomplished through modelling of a conceptual deep mine.

Role of Ventilation Air

Ventilation air is circulated through mines to dilute pollutant concentrations to safe levels and to remove them from the mine. In gold mines the main air pollutants are heat, dust, noxious and explosive gases and ionizing radiation.

As mining depths increase, the capacity of the ventilation air to remove heat from underground progressively decreases and the temperature of the ventilation air supplied from surface will eventually exceed the desired reject air temperature from the workings. As a consequence, heat produced below this depth can only be removed by other means usually refrigeration.

The specific air quantity is defined as the quantity of downcast air per 1,000 tons mined. For South African gold mines this has steadily increased with time and, in 1994 was approximately 6.0 kg/s per 1,000 tons mined per month. The corresponding mean weighted rockbreaking depth in 1994 was 1,777 m (Denysschen,1994). The increasing trend can be ascribed to reduced production as some areas in mines became unprofitable and total air quantities remained constant. However, the increase in the specific air quantity can also be attributed to the need to distribute cooling in order to remove the increased mine heat load.

Controlled Recirculation

Controlled recirculation of ventilation air involves mixing some of the reject air from a section with fresh air before reintroducing it into the section. In Figure 1, intake air is introduced into a section at point 1. It passes through the working area and is rejected back to point 6. Recirculation takes place along the route 2-3-4-5-2. The quantity of air recirculated is independent of the quantity of fresh air entering the section and can be considered as the forced circulation of the air in the circuit 2-3-4-5-2.

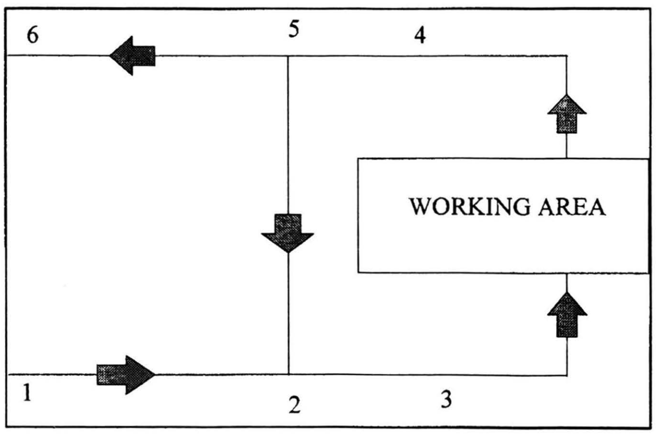

Figure 1. Simple controlled recirculation system.

The Potential Benefits of Controlled Recirculation

Considering Figure 1, superimposing controlled recirculation onto an existing ventilation system will increase the ventilation air quantity at points 3 and 4, which will result in the following:

1. The distribution of cooling within the area being more effectively undertaken: In deep mines where air flow quantities are likely to be restricted and horizontal distances to the working areas significant, up to 3,000 m, air temperatures will increase rapidly for a given heat load. With the increased air quantities provided by controlled recirculation, the rate of temperature rise will be decreased. In this context, controlled recirculation can only be considered as a means of more effectively distributing cooling and not a means of generating cooling. The amount of refrigeration required in an area is virtually independent of any controlled recirculation within that area.
2. For the same downcast air quantities air velocities will increase throughout the working area, with two beneficial effects: Increased values of cooling power without any changes in air temperature, and decreased potential gas layering. (Bakke, 1964) (Leach, 1966).
3. Reduced power costs: Smaller main ventilation fans, and thus less electrical power consumed by the main ventilation fans.
4. Less refrigeration capacity is needed to counter the effects of autocompression.

Further effects of controlled recirculation. In addition to the points listed above, controlled recirculation will affect the following:

1. Heat rejection capacity: Since controlled recirculation decreases the quantity of return air, it reduces the heat rejection capacity available for underground refrigeration plants. Cooling generation systems under consideration for deep mines include combinations of underground and surface cooling plants and the operation of water vapour refrigeration plants underground. If these developments are pursued then the availability of heat rejection capacity will remain an important issue.
2. Radiation: With the exception of radon progeny concentrations, controlled recirculation does not increase the total pollutant levels measured in the return from a recirculation area, provided that the fresh air quantity remains constant (Burton, 1984), although the average levels measured within the area do increase. The radon progeny concentration of the outlet air will increase with the introduction of controlled recirculation. There is no recognized safe lower limit for radon progeny concentrations, and any increase in average levels will increase the health risk from ionizing radiation.
3. Safety arrangements: In terms of safety, two main hazards exist: A gradual build up of contaminants, including heat, blasting fumes and dusts, if the intake air supply is interrupted. Due to the high rock temperatures in a deep mine, air temperatures will rise rapidly when cooling is interrupted. Airflow should be monitored, and in the event of failure all mining activities should cease until the intake air supply has been restored. Fire introducing fumes and smoke into the mixed intake airway. The risks associated with these failures occurring in a recirculation loop can be effectively minimized and controlled by incorporating a detection system in the recirculation loop and ensuring that recirculation is stopped once a hazard has been detected.

Controlled recirculation on South African gold mines. Prior to 1982 two controlled recirculation schemes had been documented on South Africa gold mines. These were at East

Rand Proprietary Mine in 1948, when a portion of the upcast air was cooled and mixed with the downcast air (Gorges, 1952), and in 1973 Holding reported on a controlled recirculation scheme in a single stope (Holding 1973).

Table 1. Operational and proposed controlled recirculation sites (1988).

Mine	Fresh air (m^3/s)	Rec. air (m^3/s)	Depth* (m)	VRT** (°C)
Winkelhaak	120	60	1,131	34.3
East Driefontein	65	30	1,871	39.2
Hartebeestfontein	18	24	2,220	45.5
Loraine 1#	26	45	1,699	45.5
Loraine 3#	14	27	1,699	45.5
	67	65		
	57	78		
	65	65		
	26	45		
Proposed sites				
President Steyn	150	50	1,789	45.7
Western Holdings	100	110	1,287	38.8

* Average weighted rock breaking depth
** Corresponding virgin rock temperature

During 1982 a large scale field trial was comm-issioned in the Eldorado section of Loraine Gold Mines Ltd. (Burton 1984). The scheme mixed 35 m^3/s of recirculated air with 15 m^3/s of fresh air at a depth of approximately 1,800 m. The success of this work led to the introduction of a number of controlled recirculation schemes and by 1988 nine schemes were operating on four mines in South Africa with no major operational problems being reported. These sites, together with additional proposed sites, at that time, are listed in Table 1.

At present, a scheme at Oryx mine (Rose, 1992) is operational on a South African gold mine. 300 kg/s is recirculated and mixes with 600 kg/s of fresh downcast air at a depth of approximately 2,200 m. In respect of performance and a buildup in measured pollutant levels no problems have been encountered to date.

Controlled recirculation schemes implemented on South African gold mines have either been introduced at comparatively shallow working depths or have been local in nature, handling small air quantities. To consider the potential effect that implementing a large scale scheme would have on the environmental conditions for a deep South African gold mine, a comparative evaluation was undertaken as part of a broader conceptual deep mine study.

CONCEPTUAL DEEP MINE MODEL

Model Input Considerations

To ascertain the implications of superimposing controlled recirculation for the life of a proposed mine, two scenarios were considered. The mine was initially modelled at the stage when full production had just begun, i.e. mining eight raise lines while the distance to the sub shaft was a minimum. The model was then revised to reflect the condition when both production and raise line distances from the sub shaft were at the maximum.

The study considered the most common South African gold bearing reef occurrences, namely Witwatersrand type deposits. Generally, these narrow reefs occur in widths of between 1.0 and 1.8 m, and usually dip at 20 to 40° from the horizontal.

Stope Layout and Mining Sequence

A stope layout and mining sequence suited to the concentrated mining concept was designed to achieve optimal mining conditions for depths between 3,000 and 4,500 m for the orebody type described above. This is essentially a scattered mining layout having six 40 m long panels per stope, giving an overall back length of 240 m to 250 m. Mining is carried out on one side of the centre-gully raise at a time.

The overall layout is such that development is always ahead of stoping by at least one completed raise connection, these being spaced at 250 m intervals on strike. Stoping follows the development in a progressive and orderly manner. Three levels will be connected via a raise-bored holing about 200 m from the shaft. This will allow channelling of the return air from the lower development ends to the return airways, and can subsequently be used as a dedicated controlled recirculation airway. Development of the first three levels will be done on a multi-blast (two blasts per day) basis until the section is at full production.

Factors

Factors influencing the choice and design of the underground environmental control facilities were:

- concentrated, sequential grid mining method*
- reef dipping at 20 °
- production rate of 260 kton/month (200 kton reef & 60 kton waste)
- maximum stope face advance of 25 m/month (single sided)
- average stoping width of 1.4 m

- 70 percent backfilling of mined areas
- maximum distance from sub-shaft of 3,000 m
- depth below surface between 3,000 m and 4,500 m
- multiple vertical shaft system (Main shaft plus sub-shafts)

* Sequential grid mining is where raise lines are developed up front at approximately 200 m intervals and mining then takes place from a raise line, in both directions, to approximately halfway to the next raise line (the 'stop' line).

Design Reject Wet Bulb Temperature and Air Velocities. The design reject wet bulb temperature was set at 27.5 °C for stopes and development end faces. The maximum wet bulb temperature for all other areas such as development end back areas, haulages and workshops, was set at 29.0 °C. It was assumed that work rates in these areas are lower and may be assisted by machinery and that air flow velocities (and thus cooling power) are generally much higher than in stopes and development ends.

The minimum design air velocity at a stope face was set at 1.0 m/s. The gain in cooling power with air velocities higher than this does not in itself justify the required increase in fan power. A higher air velocity in a stope results in a greater pressure drop across the stope, which increases the fan power requirement.

The high virgin rock temperature (about 60°C in this model) is by far the greatest contributor of heat, particularly in stopes, where a large fresh rock surface area is exposed daily. This results in a rapid rise in air temperature if the air traverses the stope too slowly. Apart from cooling the air, the effective temperature rise of the air can be curtailed by ensuring that a sufficient quantity of air is available to "dilute" the effect of the added heat.

The maximum air velocity in tunnels was set at 6.0 m/s since the increase in pressure drop beyond this point does not justify the required increase in fan power to overcome the resistance to airflow. The minimum quantity of air supplied to a development end face, with a forced/exhaust system, was set at 0.5 m^3/s per m^2 of face. This will result in an average face velocity of about 1.0 m/s, and a back area velocity of about 0.5 m/s.

Controlled Recirculation of the Ventilation Air. The design parameters set for the implementation of a controlled recirculation system were: fifty per cent of the air used to ventilate stopes and developments would be recirculated and reconditioned. This fraction would amount to a global recirculation figure of 40 per cent, taking into account the effects of air leakage.

For the successful implementation of such a system, both the siting of the recirculation pathways and the sizing of the recirculation fans were considered critical. In this particular instance, it was deemed important, from a practical point of view, to excavate one dedicated recirculation raise on either side of the sub-shaft between the lower main intake and the main return level. These raises would serve for the entire period that mining operations were in progress between the levels. Thus, the recirculation installations would only have to be re-sited when mining operations began on the next lower level.

Fans. In terms of fan installation and maintenance, the prime consideration was to avoid wherever possible the necessity for underground fans. In the case of once-through ventilation, the use of underground booster fans would therefore be avoided if possible, so that a single surface fan installation would handle the total ventilation requirement of the mine. In the case of recirculation, the number of underground fans should be limited only to those necessary for the controlled recirculation system.

Cooling Strategy. Due to its high latent heat capacity, ice is currently considered to be the most suitable cooling medium for deep mines. Ice production occurs on surface, and it is assumed that an 85 per cent ice /15 per cent water slurry at 0°C is the primary source of cooling in the model mine under consideration. In terms of the global ventilation layout, the main shaft from surface to the transfer level is assumed to be a bratticed shaft.

The total amount of heat contributed to the ventilating air by the surrounding rock is to a large extent dependant on the temperature difference between the air and the rock surfaces. By keeping this temperature difference as small as possible within the limits of the design specifications, the total heat flow rate from the exposed rock surfaces can be substantially reduced. Despite the increased capital cost for the cooling equipment involved, such as piping, pumping and pipe insulation, particularly as distances increase, the general principle applied to the model was distributed localized cooling.

Within the model consideration had to be given to mid-shaft cooling (at the transfer level) to overcome the effects of autocompression, and to ensure that the temperature of the air reaching the sub-shaft and its stations was within the design limits.

The strategy relating to underground bulk air coolers was to limit their use as far as practicable, in keeping with the overall principle of distributed localized cooling. In terms of this strategy, the purpose of bulk air cooling would be to ensure that the air temperature in main intake airways remains within the design limits up to the point where the air enters the working areas. The air for the working areas would be conditioned at the intakes to the stopes and development ends.

Chilled service water provided additional cooling in the working areas, augmenting the air coolers at the intakes. The design values with regard to chilled service water are that the water temperature will reach the working areas at 15.0 °C and is used at the rate of one ton of water per ton of rock broken.

Environmental Simulation. In order to assess the underground environment of a deep mine in which the preceding strategies would be implemented, two software tools, namely ENVIRON 2.5 (CSIR Mining Technology) and, in the case of recirculation, RADCIRC (CSIR Mining Technology), were used to simulate the different scenarios. A base-case model was first constructed to simulate a once-through situation (i.e. no recirculation of air). Regulators and coolers were then added in order to obtain the design environmental conditions. Subsequently, recirculation loops were added, and the regulators and coolers adjusted to again yield the design environmental specifications.

Initially, worst-case scenarios in terms of air heat loads were assumed. Thus, the maximum face advance rate in stopes and development ends, the maximum distance of working areas from the shaft(s), the absence of tunnel insulation and additional linear heat loads in all tunnels were included. The reason for these assumptions is so that the maximum cooling requirement of each cooler could be determined. Thereafter, the assumed production rates were relaxed to the design values in order to determine the actual overall cooling power requirement.

Some simplifying assumptions were included in the models. The proposed symmetry of mining operations to either side of the sub-shaft suggested that only one half of the mine needed to be fully modelled. Provided that the other half could be adequately represented in the network by a simple sub-network in which air quantities and conditions would equal those in the fully modelled half. Such a sub-network was constructed from three resistances with fixed air temperatures at the return node points. It was also considered important to make allowance for inefficiencies in airflow control measures. Therefore, the models included branches, which allow a portion of the fresh air to leak directly from the main intakes to the main return. Figure 2 shows the network used to model the controlled recirculation configuration.

Table 3 shows a comparison of power requirements of the two ventilation strategies, namely with and without recirculation. For the purpose of calculating ice quantities and cooling figures, the reject water temperature was assumed to be 27.5 °C in all cases. Under these conditions, the once-through scenarios required 162 and 189 kg/s of ice and the controlled recirculation scenarios 153 and 189 kg/s. The coefficient of performance of the refrigeration plant was taken as 5.5 and that of the ice plant as 4.0.

Table 4 presents a comparison of annual capital and running costs in US $ M between the two ventilation and cooling systems. The annual capital cost is equivalent to the total capital shared over 10 years and assumes an interest-free amortization period. The current annual cost of electrical power is assumed to be US $ 180,000 per MW.

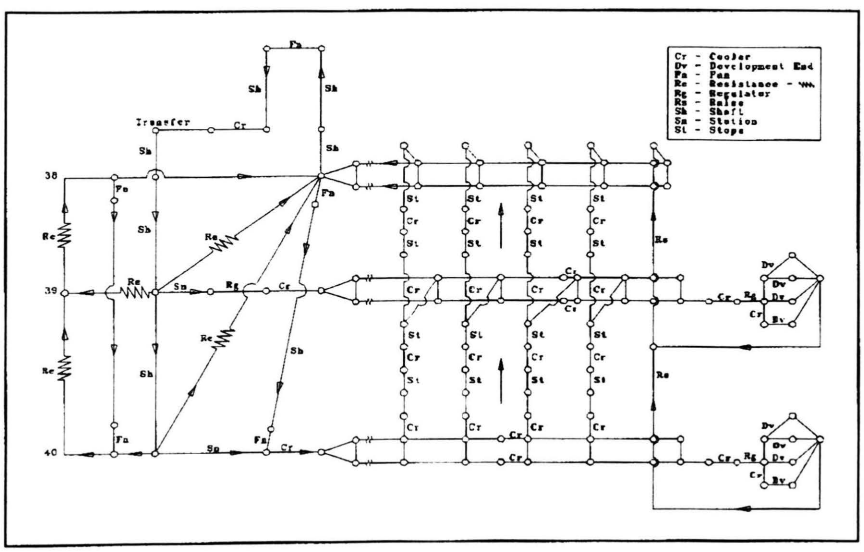

Figure 2. ENVIRON network (with recirculation) used in the simulations.

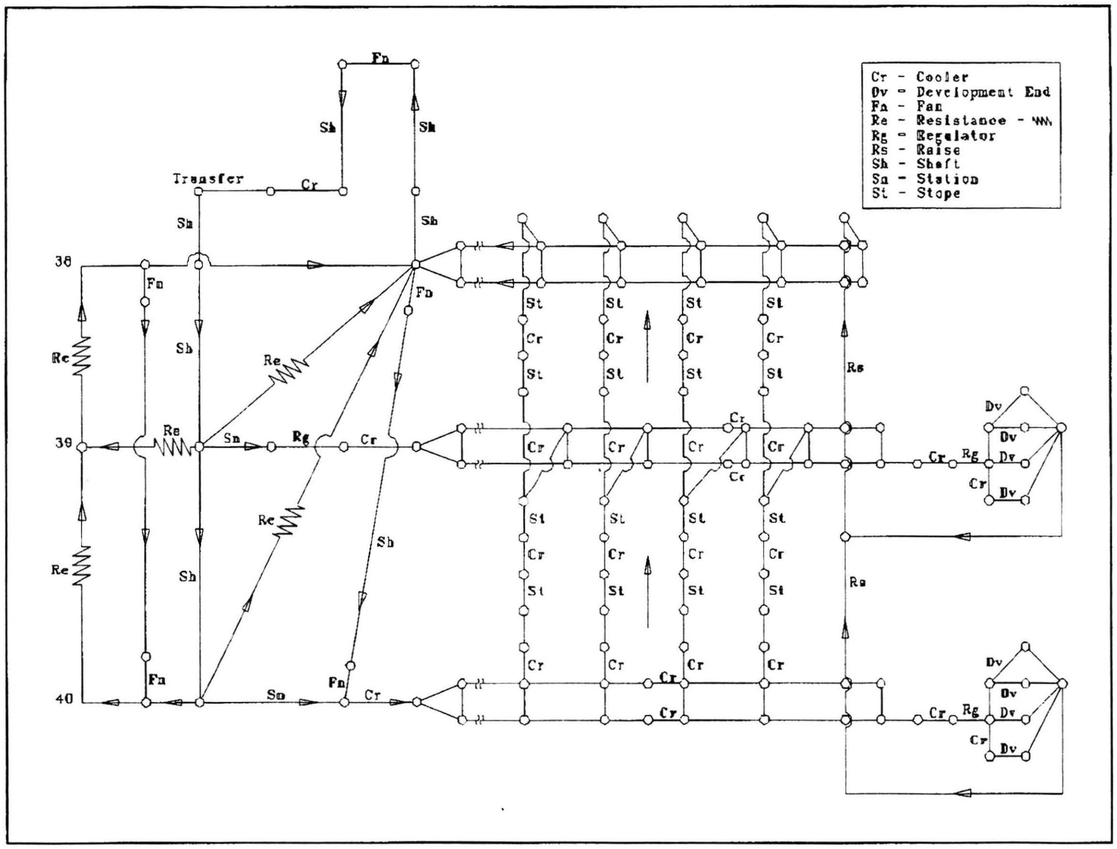

Figure 3. ENVIRON network (with recirculation) used in the simulations.

Simulation Results

The cooling and fan power results are summarized in Table 2 below.

Table 2. Summary of cooling capacity and fan power results.

	Start of full mining cycle		End of full mining cycle	
Item	Without Re-circulation (kW)	With Recircu-lation (kW)	Without Re-circulation (kW)	With Recircu-lation (kW)
Cooling:				
a) Mid-shaft cooling	9,000	5,000	9,000	5,000
b) Bulk air	3,000	4,000	12,000	16,000
c) Tertiary cooling	33,000	33,000	33,000	33,000
d) Distribution losses	9,000	9,000	9,000	9,000
Total installed rated cooler capacity	54,000	51,000	63,000	63,000
d) Chilled service water	8,000	8,000	8,000	8,000
Fan Power:				
a) Main surface installation	6,800	1,500	6,800	1,500
b) Booster fans	N/A	800	N/A	800
Total fan power	6,800	2,300	6,800	2,300

Table 3. Comparison of cooling duties and electrical power requirements

	Start of full mining cycle		End of full mining cycle	
	Without Recirculation	With Recirculation	Without Recirculation	With Recirculation
	Electrical power used (MW)	Electrical power used (MW)	Electrical power used (MW)	Electrical power used (MW)
Surface refrigeration plant	2.2	2.1	2.6	2.6
Surface ice plant	11.0	10.3	12.8	12.8
Pumping power	6.3	5.9	7.6	7.6
Fan power	6.8	2.3	6.8	2.3
Total power	26.3	20.6	29.8	25.3

Table 4. Comparison of annual capital and overall costs (US $ M).

	Start of full mining cycle		End of full mining cycle	
	Without Recirculation	With Recirculation	Without Recirculation	With Recirculation
Surface refrigeration plant	0.5	0.5	0.5	0.5
Surface ice plant	15.8	15.8	15.8	15.8
Underground coolers	0.5	0.5	0.5	0.5
Pumps	2.5	2.5	2.5	2.5
Fans	0.3	0.1	0.3	0.1
Raise-boring	N/A	0.1	N/A	0.1
Capital	19.6	19.5	19.6	19.5
Power	4.7	3.7	5.4	4.6
Total	24.3	23.2	25.0	24.1

Discussion of Simulation and Results. Table 2 shows that the bulk air cooling requirement increases in the case where controlled recirculation is practised. The reason for this is that the heat added to the air in the main returns and the recirculation airways, in conjunction with that added by the recirculation fans, approaches the heat of autocompression in the once-through ventilation system. Bulk air coolers, near the sub-shaft in the lower main intakes remove the additional heat, which is reflected in Table 2 as a difference in the respective bulk air cooling requirements for the two systems. Predictably, the mid-shaft cooling requirement in the recirculation system is substantially lower than for the once-through system. The reason for this is that the greater quantity of downcast air in the once-through system picks up additional heat due to auto-compression in the shaft. The total quantity of air (on surface) in the once-through scenario is 582 kg/s and 301 kg/s in the case of recirculation.

The results have indicated that the downcast air quantity to monthly production ratio for this mine is 2.23 m^3/s per kton per month in the once-through ventilation system, and 1.13 m^3/s per kton per month in the controlled recirculation system (at a density of 1.0 kg/m^3). The low specific air quantity can be explained by the inclusion of effective backfill placement in both models resulting in minimum leakage into back areas. The installed cooling to monthly production ratio for the model mine under consideration amounts to 204 kW and 238 kW per kton per month in the once-through strategy, and to 192 kW and 238 kW per kton per month in the recirculation strategy.

In the once-through ventilation system, the high air quantities and low total residence time in the mine suggest that exposure to radiation should be low. This is borne out by RADCIRC simulations of the recirculation system, where similar air quantities are passed through the stopes, but the residence time of the air in the mine is much higher. In the latter case, the RADCIRC simulations indicated that average radiation exposure would not be a significant problem, even with a high radiation source term (approximately 1 kBq/m^3). To implement controlled recirculation broadly across the South African gold mining industry, a method to reduce the radiation level in the recirculated fraction of the ventilation air, to a level at or near that of the fresh intake air, will need to be investigated. While it is not the intention to advocate any particular method for reducing the radiation hazard on mines, it is important to recognize that solutions may exist. At least two options have or are being considered. These are either to scrub the recirculated ventilation air fraction prior to re-mixing with the fresh intake air, or to cyclically operate the recirculation fans, thereby flushing the system at regular or convenient intervals.

CONCLUSION

By incorporating the chosen strategy, an effective controlled recirculation fraction of 42 per cent was used in the simulations. The expected reductions in the heat load from autocompression and fan power requirements were found. In the case of fan power requirements, a reduction of 66 per cent was predicted.

Despite the specific nature of the mine model, controlled recirculation proved to be the most economical option at the start of and at the end of the mining life cycle. Overall savings through the introduction of controlled recirculation were significant, particularly in terms of energy consumed. The power saving at the beginning of the mining cycle amounted to 5.7 MW and at the end of the cycle 4.5 MW with corresponding cost savings of US $ 1.0 M and US $ 0.8 M per annum respectively. The variation in the predicted savings between the two stages in the mine cycle can be explained by the longer return airway distances, at the end of the mining life, adding heat to the recirculating air fraction.

The overall cost savings, including the annual capital cost component, at the beginning of and at the end of the mining cycle were US $ 1.1 M and US $ 0.9 M. Although this represents a reduction in the total cooling cost of only 4.6 and 3.7 per cent respectively, the benefits of controlled recirculation will inevitably become more apparent as mines become deeper and hotter. While it is possible that in some instances existing infrastructure may prevent present gold mines from realizing certain benefits of controlled recirculation, in most cases when planning a new deep mine or section, infrastructure that permits the implementation of controlled recirculation should be considered.

Notwithstanding the financial considerations, for many deep South African gold mines when there is a shortage of ventilation air, it may be necessary to choose between sequentially reusing ventilation air or using controlled recirculation. In terms of radiation, the ALARA (As Low As Reasonably Achievable) principle would favour the system that leads to the lower radiation levels. If measures were put in place which either reduce or control the radon progeny concentrations in the recirculated fraction, then the preferred option could well be controlled recirculation.

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